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MILLING METHODS AT THE MIDVALE CONCENTRATOR OF THE U. S. SMELTING, REFINING & MINING CO., MIDVALE, UTAH

BY

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By R. A. Pallanch[2]

INTRODUCTION

This paper, which describes the milling practice at the Midvale concentrator, is one of a series being prepared by the Bureau of Mines.

The Midvale concentrator, a custom lead-zinc selective flotation plant, is in Salt Lake County, Utah, on a flat site several hundred yards east of the Jordan River and about 200 yards south of the Denver & Rio Grande Western Railroad tracks leading to Bingham Canyon. The site is at an elevation of about 50 feet above the river, the highest available location, thus allowing maximum storage of tailings.

The concentrator operates as an all-flotation plant, which means that all separations of the different concentrates and tailings are effected by flotation methods. A very insignificant amount of pyrite is recovered by tables from the final flotation tailings, but this is incidental, as will be explained later.

The concentrating plant consists, in the main, of three steel-constructed buildings housing, respectively, the crushing equipment, the storage bins and grinding, concentrating, and filtering units, and the thickener equipment. They are referred to locally as the crushing plant, the flotation mill, and the thickener building. The crushing plant is connected to the flotation mill at a point above the storage bins by an inclined conveyor gallery. A manway, forming also a pipe gallery, connects the flotation mill and the thickener building.

Water for concentrator operation is supplied by a canal which receives water from Utah Lake. Power is supplied by the Utah Power & Light Co. It is brought to the mill at 44,000 volts and is transformed to 440 volts for plant use.

ORE SUPPLY

Most of the mill ore supply originates in underground mines of the Bingham district. Two mines at Bingham and now operated by the U. S. Smelting, Refining & Mining Co.—the U. S. mine and the Lark mines, the latter formerly known as the Bingham mines—supply about 750 tons of ore per day. Of the total tonnage furnished by these two mines the U. S.

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6492."

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mine ships a regular daily tonnage of ore amounting to between 400 and 450 tons, locally referred to as "U. S. crude." In addition to the regular daily tonnage of company ore from this mine daily shipments amounting to between 100 and 150 tons are made by lessees from widely scattered portions of the same mine. The Lark mines ship an average of more than 200 tons of ore daily. Besides these company-controlled ores considerable custom ore finds its way to the mill from other lead-zinc-pyrite underground workings of the Bingham district. Small amounts are also received from Park City and Tintic, together with some from the neighboring States of Colorado, Idaho, and Nevada. The total ore received provides the mill with 900 to 1,000 tons daily. The present daily capacity of the plant is slightly more than 1,000 tons. Any shortage of ore is usually compensated by the shutting down of a mill section rather than by cutting the rate of feed to all sections.

CHARACTERISTICS OF ORES TREATED

The ores handled in the concentrator vary widely in metal content, physical characteristics, degrees of hardness, fineness and oxidation, gangue constituents, and soluble salts. In general, it may be stated that the mill now handles principally ores containing large amounts of pyrite as compared to both galena and sphalerite and negligible amounts of copper minerals. Much of the ore handled has contained more than 50 per cent of pyrite and less than 10 per cent of sphalerite.

To make the description of ores handled more specific, it will be necessary to confine mention to the "U. S. crude" ore; when this article was written (April, 1930) that ore was the main supply and formed over 40 per cent of the total mill feed. The U. S. crude ore is fairly uniform, with variations of several per cent in lead, iron, and zinc content. An average analysis is given in the following table.

Ounces per ton		Per cent						
Au	Ag	Cu	Pb	Ins.	Fe	Zn	S	CaO
0.05	4.1	0.25	9.5	40.2	12.0	9.0	19.0	1.0

The ore is fairly coarse grained, especially as regards pyrite and galena. The sphalerite is associated more closely with the gangue, making it necessary to grind about 95 per cent of the ore through 65-mesh size to obtain a satisfactory release of sphalerite. As may be judged from the assays, about 10 to 12 per cent of the total weight of the ore is galena, 25 to 30 per cent pyrite, and 12 to 14 per cent sphalerite. The balance is mostly gangue constituents, principally quartzite, with some lime carbonate, some porphyry, and considerable amounts of claylike material locally referred to as "talc."

The sphalerite is chocolate-colored; presumably the color is due to ferrous iron. The results of work on this mineral tend to the conclusion that it is partly marmatitic. Hand-picked free sphalerite crystals assay 57 per cent of zinc, 5.5 per cent of iron, and 31.4 per cent of sulphur. Cleaned zinc concentrates assay 56.5 per cent of zinc, 5.5 per cent of iron, and 32.2 per cent of sulphur. In either instance there is not enough sulphur content to allow figuring the zinc and iron as true sphalerite and pyrite, respectively. In fact, in the case of the hand-picked mineral, there is only enough sulphur to form FeS, after sufficient sulphur is allowed for the zinc to be present as ZnS.

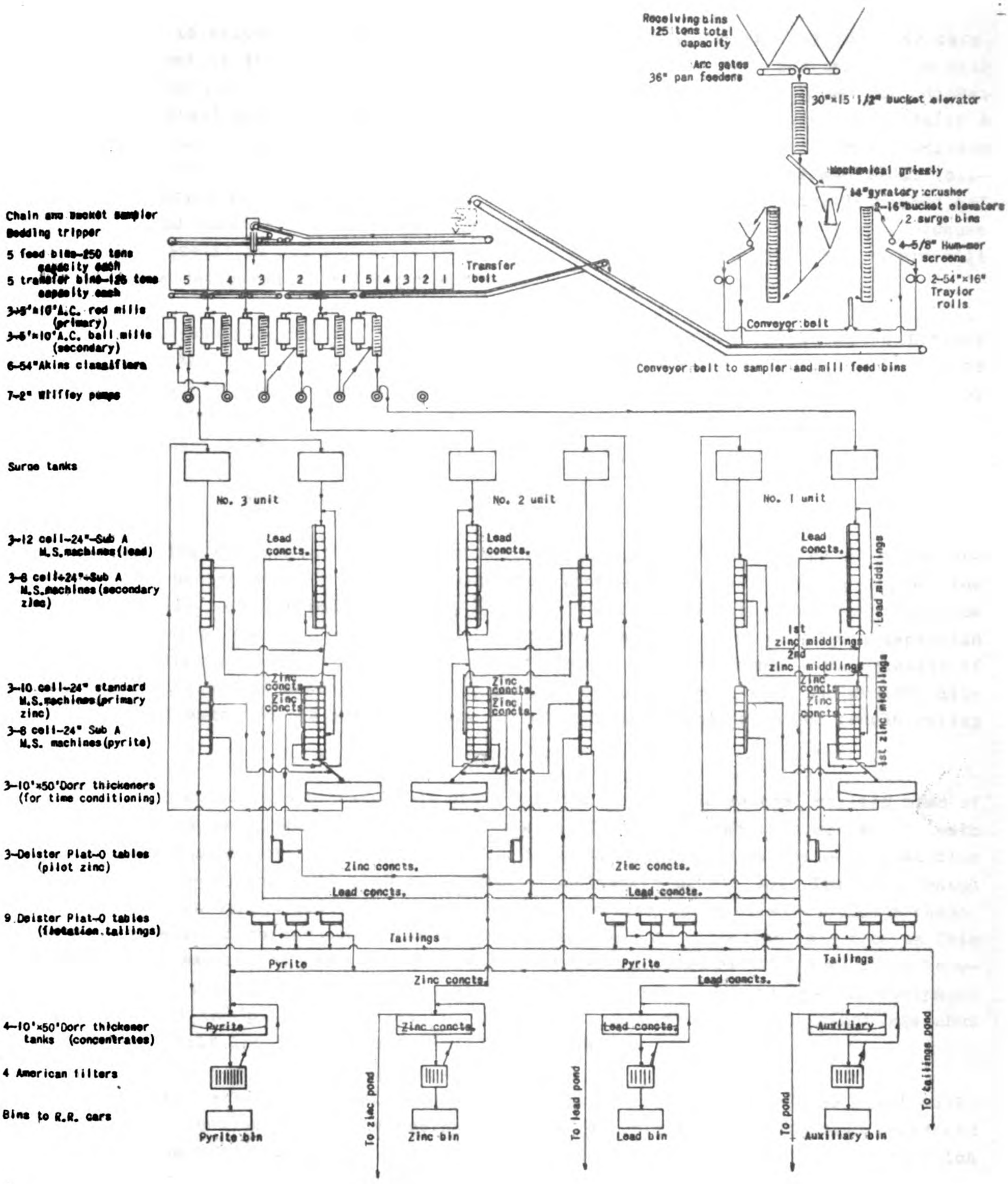

Figure 1.- Flotation mill flow sheet

The ore is shipped from Bingham in open all-steel 50-ton-capacity railroad cars. The moisture content in the ore, as received, ranges from about 5 to 7 per cent. Ore with 5 per cent of moisture is readily handled; that containing 7 per cent is very wet and sticky, causing trouble at all points in the crushing, conveying, and feeding systems. Usually a very minor portion of the galena in the ore is sulphatized, the variations of this condition bringing about variations in the lead content of the lead-circuit tailings and final tailings. The presumption is that this tarnished material on the galena surfaces is washed off during grinding and becomes a very inpalpable material of the finest micron size. Because of its very fine size it is not recovered by gravity methods and its sulphatized nature will not allow it to be recovered by any known economic flotation treatment.

A screen analysis of the dry-crushed material, as introduced into the primary grinding mills, is included in Table 1. The amount of minus 200-mesh material shown in this table (12.6 per cent) is quite characteristic of the U. S. crude ore and is accounted for by the talcy material previously referred to, which, upon wetting, immediately disintegrates and passes into the finest size.

BRIEF HISTORY OF CONCENTRATOR OPERATIONS

The milling of ores at Midvale, Utah, commenced with the building of a 90-ton-capacity gravity concentrator in 1905. Later additions at intervals finally raised the capacity of this mill to about 465 tons daily in 1915. Jigs and tables were used to produce lead-iron and zinc-iron concentrates. The zinc-iron concentrates were dried and separated into zinc concentrates and iron concentrates by Huff electrostatic machines. The capacity of operations was maintained at about 465 tons daily until early in 1926, when the present all-flotation treatment absorbed the available ore and caused abandonment of other concentrating methods.

From 1915 until 1923 considerable flotation testing of the several milling ores of the company was conducted. Early in 1924 this work finally culminated in a series of tests which conclusively checked the all-flotation work which had been done on United States mine and custom ores. The building of a 25-ton-capacity continuous pilot flotation unit, which commenced operation in April, 1924, resulted. Besides checking up on all-flotation methods, comparative tests which incorporated table concentration ahead of flotation were made in this plant. The gravity section of the pilot mill was later discontinued as it proved of no benefit and complicated the flow sheet to a great degree. In early 1925 additional equipment installations raised the pilot-mill capacity to over 60 tons daily. This plant was shut down after the new mill had treated accumulated ore stocks.

The present mill flow sheet is based largely on the findings of the pilot mill. The buildings were constructed and the equipment installed during 1925. The mill commenced operations on February 24, 1926, the rated capacity being 750 tons daily. The installation of a sixth grinding mill in December, 1926, brought the rated capacity to about 850 tons; the mill is now handling more than 1,000 tons per 24 hours.

PRESENT CONCENTRATOR PRACTICE

Figures 1 and 2 present the flow sheet of treatment and a section of the concentrator building, respectively.

Unloading of Ores

A railroad trestle runs alongside of and beyond the crushing plant; space is provided for the spotting of eight full-size steel standard-gage drop-bottom cars at one time. Two or three settings of the cars are made during the day shift as needed. The ore is dumped into two steep-sloped self-cleaning hopper-bottomed bins of about 75 tons capacity each. Directly beneath these receiving bins are two continuous 36-inch manganese-steel apron feeders which travel toward each other. They are driven by chain and sprocket drives from spur-gear reducers and 7½-hp. motors. The rate of ore feed is adjusted by means of arc gates The combined load of the two feeders amounts to about 125 tons per hour. The crushing plant operates only 8 hours of the day, barring lost time, to supply the needed 1,000 tons of ore to the storage bins.

The arrangement of receiving bins and feeders as described allows the mixing of two separate ores in the crushing plant without resorting to bedding. This practice has proved of inestimable value.

Coarse Crushing

The two apron feeders deliver ore to a continuous chain and sprocket elevator equipped with 30 by 15½ inch steel buckets and manganese-steel links. The elevator is driven by a 20-hp. motor through a worm-gear reducer connected to the head pulley and discharges its contents onto a 36-inch standard Nelson mechanical grizzly. The latter is operated by a chain-and-sprocket drive from the shaft that drives the elevator and therefore starts and stops with the elevator. The grizzly is equipped with nine manganese-steel bars set with 2-inch spaces which are kept clear by the tips of manganese-steel fingers passing through at intervals.

The grizzly oversize passes to the mouth of a 14-inch Traylor bulldog gyratory breaker set with an average discharge opening of about 2 inches. The breaker has manganese-steel concaves and mantles which last more than a year when used 8 hours per day, and when discarded only a small fraction of their total weight has been worn off. The breaker is connected by belt to a 75-hp. motor.

Intermediate Crushing

The undersize product of the mechanical grizzly and the breaker product join and are elevated by a 16-inch rubber-belt elevator equipped with 16 by 9 inch malleable-iron buckets to a hopper which feeds two 4 by 5 foot No. 33 Hum-mer heavy-duty vibrating screens. The latter have ¾-inch square openings except when screening very wet ore, when ⅜-inch Ty-rod screens are substituted. A fairly large percentage of the ore, probably amounting to 40 per cent of the feed weight, passes through these screens, and the undersize to a large extent is made up of fines which occur in the run-of-mine ore.

The oversize of these screens passes to a feed belt, over which an electromagnet is suspended to remove tramp iron. This belt discharges the ore to a set of 54 by 16 inch Traylor rolls set about 1 inch apart. The roll product is elevated by another 16-inch rubber-belt elevator equipped with 16 by 9 inch malleable-iron buckets, to a hopper which feeds two 4 by 5 foot Hum-mer heavy-duty screens having ⅝-inch square opening. These screens operate in closed circuit with a second set of 54 by 16 inch Traylor rolls set ⅜ inch apart.

The rolls crush the screen oversize, which is returned to the same elevator that handles the first roll product. A second feed belt over which a second set of electromagnets is suspended conveys the circulation-screen oversize, which totals two to three times the amount of undersize, to the second set of rolls. Unlike the first-screen undersize the undersize of the second set of screens is composed mostly of sharp, angular quartzite particles, there being but a small amount of primary fines present. The two screen undersize products join and comprise the final product of the crushing plant.

The two sets of 54 by 16 inch rolls are driven by straight and cross belts at a speed of about 95 r.p.m. from two separate line shafts direct-connected to 75-hp. motors. The roll shells are made of cast steel, and each weighs 3,400 pounds when new and about 500 pounds when discarded. The shells of the primary set of rolls, which operates in open circuit, show a consumption of steel amounting to 0.019 pound per ton of ore passing through the crushing plant. The shells of the secondary set, which operates in closed circuit with the ⅝-inch screens, show a consumption of 0.025 pound of steel per ton of ore. The total steel loss due to wear of roll shells in the plant is therefore 0.044 pound per ton of ore milled.

All hoppers and ore chutes in the crushing plant are lined with white cast-iron liners about ⅜ inch thick, which are readily replaced when worn. The life of screens averages from forty to sixty 8-hour days of operating service.

CONVEYING, SAMPLING, AND STORAGE OF CRUSHED ORE

The combined undersize products of the two sets of screens in the crushing plant are carried to an automatic chain-and-bucket sampler by a conveyor belt. At this point the U. S. and Lark mine ores are sampled in eight and six car lots, respectively.

The chain-and-bucket automatic sampler consists of two parallel chain-and-sprocket systems about 18 inches apart placed at right angles to the line of the inclined belt conveyor. The top of the sampling system is just below the head pulley of the conveyor. The conveyor discharges its load vertically between the chains, which carry two steel buckets about 3 inches wide and 9 inches deep. These buckets cut the ore stream at right angles at regular intervals and deposit their loads into a small hopper, which feeds a revolving horizontal disk. The latter is discharged by a scraper at a point directly above a turn-back in the chain system. Each of the two buckets, therefore, makes a cut of the sample as it discharges from the horizontal disk, and the final sample is deposited in one of two hoppers reserved for this purpose. As the ratio, by weight, of the first sample and the total ore is about 1 to 50, the ratio of the final sample to the total ore is 1 to 2,500, which means that a sample weighing about 40 pounds is obtained from a carload of ore. From the storage hopper the sample is crushed to minus 4-mesh size by a small set of rolls, after which it is again cut by a Snyder sampler and further prepared for the assay office by pulverizer and riffles.

All ores except U. S. and Lark are shipped to the mill in one or two car lots and the sampling of these smaller lots by the method described would delay and seriously impede crushing-plant operations. For this reason all such lots of ore are usually sampled at the Midvale smelter sample plant, and occasional lots are sampled elsewhere.

After the ore passes through the chain-and-bucket automatic sampling system it goes to a 16-inch horizontal belt conveyor, which runs directly above the center line of a row of steel bins of inverted parabola shape in cross section. The conveyor is equipped with an automatic tripper, which permits discharge of ore into any bin or series of bins.

The bin arrangement of the concentrator is unique in that it provides for separate storage of ores or mixtures of ores. There are five main ore-storage bins, each of 250 tons capacity and each equipped with three roll feeders, which discharge onto collecting conveyor belts which in turn discharge onto conveyors which feed the various grinding mills.

In addition to these main bins there are five so-called transfer bins, each of 125 tons capacity and each equipped with one roll feeder. Referring to Figure 1, the transfer of ore from these bins to the main ore bins is effected as follows: The roll feeders of the transfer bins discharge onto an inclined conveyor belt, which is 16 inches wide and operates at a speed of 100 feet per minute. This conveyor in turn elevates the ore and discharges it onto the conveyor, which carries ore from the crushing plant to the main ore bins. The transfer is made at a point about midway between the crushing plant and the sampler.

With the arrangements described it is possible to mix ores at this plant by the following three methods:

1. As previously stated, a mixture of two ores may be made in the crushing plant by discharging the two receiving bins, which contain different ores, onto feeder belts which travel toward each other and discharge at a common point.

2. Ore can be discharged from any or all transfer bins and this ore can be fed on top of any ore or ore mixture en route from the crushing plant to the ore bins.

3. By discharging ores from two main ore bins and feeding these to the same primary grinding mill of a mill section, mixtures of ores or of ore mixtures may be introduced into any mill section.

Often a combination of all three methods of ore mixing is used to accomplish the desired result of balancing ore feeds to any mill section.

GRINDING

Starting with grinding operations and ending with flotation treatment the mill is divided into three sections. After flotation operations, like concentrates from the three sections are combined and pumped to the same thickeners for dewatering.

The grinding unit of each section comprises one 5 by 10 foot Allis Chalmers rod mill, one 5 by 10 foot Allis Chalmers ball mill and two 54-inch partly submerged Akins classifiers.

The rod mill at present receives 350 tons of ore per 24 hours from the storage bins and operates in closed circuit with one of the Akins classifiers. The classifier sand amounts to approximately 200 tons per day and is returned to the rod mill; the overflow contains about 55 per cent of solids and is elevated by a 2-inch centrifugal sand pump to the feed box of the second Akins classifier. In this classifier the pulp is diluted with water to about 32 per cent of solids, which permits practically all of the material that requires additional grinding to settle, the sand product comprising the feed to the ball mill. The ground ball-mill product is returned to the same classifier; the ball-mill circulating load amounts to about 500 tons per 24 hours.

The rod mills and ball mills operate at speeds of about 18 and 27 r.p.m., respectively, and each mill is driven by a 100-hp. induction motor, direct connected by means of herringbone gear and pinion, flexible coupling, and spur-gear reducer. All grinding mills are operated with pulp densities that range from 75 to 80 per cent of solids. They are lined with manganese-steel wave-type breast liners; end liners are made of either manganese or chrome steel. The rod mills are kept fairly well loaded with steel rods which contain from 0.8 to 1.0 per cent of carbon. The rod load probably exceeds 20,000 pounds, and the wear of steel is compensated by the addition of 2½-inch and 3-inch sizes, the samller sizes needed being provided by the wear of the larger sizes. The ball mills require relatively light ball loads except when the ore is unusually resistent to grinding. The normal ball charge of one of these mills probably does not amount to 15,000 pounds. Chilled cast balls, made locally, mostly 2-inch size with some 2½ inch size, are used. Hardened forged-steel balls of the same sizes show somewhat better life but have proved less economical thus far when reduced to a cost basis.

The consumption of steel per ton of initial feed to the grinding mills, when grinding U. S. ore, is approximately as follows.

	Pounds
Rod mills	
Rods	1.20
Breast liners	.19
End liners	.02
Ball mills	
Balls	1.10
Breast liners	.19
End liners	.02
Total	2.72

All Akins classifiers slope about 3 inches per foot and are driven at a speed of 5½ r.p.m. by belts from one long drive shaft. Each driving pulley is equipped with a friction clutch, which permits starting or stopping of the individual machines. The line shaft in turn is driven from a motor through a flexible coupling and spur-gear reducer. The classifier blades or flights, which are made of chilled cast iron, are extremely rigid and very resistant to wear and have an average life of more than one year. The classifier-sand products are conveyed to their respective grinding-mill feed boxes by launders set at a slope of 3½ inches per foot. The launders which convey the grinding-mill discharge products to the classifiers slope about 1½ inches per foot, and this rather flat slope necessitates the use of considerable water in these launders. All launders and classifier-overflow boxes are lined with gum rubber one-fourth inch thick.

Screen analyses of rod-mill feed and various products of the grinding stages are presented in Table 1. The figures in this table show the great reductions that, in the coarser sizes, by a single pass of the ore through the rod mill, only 0.4 per cent of the rod-mill discharge was coarser than 20-mesh. Although the rod mill reduces most of the ore to between 20-mesh and 100-mesh it produces very little additional minus 200-mesh material,

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since the 27.2 per cent minus 200-mesh material shown in this product is only 15 per cent additional to the amount shown in the feed. The figures given in Table 1 also indicate rather high classifier efficiency. The rod-mill classifier-sand product contains only 8.8 per cent of minus 200-mesh material, and the ball-mill classifier-sand product contains but 11.2 per cent of this material.

FLOTATION

The overflow product of the second Akins classifier contains 30 per cent of solids and is pumped to a 10-foot-diameter surge tank by a 2-inch Wilfley sand pump which operates with a lift of about 50 feet. From the surge tank the pulp flows by gravity, with needle-valve control, to the head of the lead-circuit flotation apparatus, a 12-cell, 24-inch Minerals Separation machine of the subaeration type. The first five to eight cells produce finished lead concentrates; the remaining cells produce lead middlings products which are returned to the head of the lead circuit by a Wilfley pump.

The tailings of the lead section are conveyed by a short launder to the head of a 10-cell 24-inch Minerals Separation standard machine, locally referred to as the "primary zinc" machine. The first three to five cells, depending upon the zinc content of the feed, produce finished zince concentrates. The remaining cells produce first zinc middlings products which are returned to the head of the zinc circuit by a Wilfley pump. The tailings of this machine are pumped to a 50-foot-diameter Dorr thickener, where they remain for 5 to 10 hour periods to condition the pulp for further flotation of zinc. Incidently, the pulp is thickened from approximately 15 per cent solids in the feed pulp to about 25 per cent solids in the thickener discharge pulp. The thickened pulp is elevated to a second 10-foot-diameter surge tank by a 2-inch Wilfley pump. The surge tank, with needle-valve control, feeds an 8-cell 24-inch Minerals Separation subaeration machine, which is locally referred to as the "secondary zinc" machine. The first two cells of this machine produce additional first zinc middlings products, which are returned to the head of the zinc circuit. The remaining six cells produce second zinc middlings products, which flow by gravity to the seventh cell of the primary zinc machine.

The tailings of the secondary zinc machine pass down a short launder to an 8-cell 24-inch Minerals Separation subaeration-type machine referred to as the "pyrite machine." Originally this term was correct, but since the concentrator capacity is increased to 1,000 tons per day the first four cells of this machine produce third zinc middlings products which join the tailings of the primary zinc machine for conditioning in the thickener. The last four cells produce pyritic concentrates, and the tailings of this machine are the final flotation tailings.

The reagents used are given in the table that follows:

Lead circuit

- Zinc depression:
  - Sodium sulphite
  - Zinc sulphate
- Lead promotion:
  - Potassium xanthate
  - Republic 19 B oil
  - Cresylic acid

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Zinc circuit

- Copper sulphate
- Republic 19 B oil
- Yarmor steam distilled pine oil
- Potassium xanthate
- Hardwood creosote

Pyrite circuit

- Fused sodium sulphide

These reagents, with the exception of copper sulphate, are introduced at various points in the flotation circuit in liquid form by Geary reagent feeders of the tandum type, with friction disks.

Each feeder is connected by a pipe line to a 6-foot-diameter storage tank of 1,000 gallons capacity. These tanks, eight in number, are supplied each day shift with sufficient amounts of solutions and oil mixtures to assure adequate supplies of reagents at all times. The level of oil or solution in each Geary feeder tank is maintained constant by means of a float valve.

All flotation machines, which number 4 to a section and 12 to the plant, are gear-driven by motors connected by means of flexible couplings and spur-gear speed reducers. The four 12-cell lead subaeration machines are operated by 60-hp. motors, the four 10-cell primary machines by 50-hp. motors, and the eight 8-cell subaeration secondary zinc machines by 40-hp. motors. The spindles rotate at 230 r.p.m., which is equivalent to slightly more than 1,400 feet per minute peripheral agitator speed when the agitators are new.

The agitators of the various flotation machines are made of white cast iron, with a soft iron core for the shaft bore. The subaeration machine closed-type agitators last 12 months on the average when used continuously. The open cruciform agitators used in the standard machines give about 15 months of continuous service. The average tonnage handled by a machine in the lead circuit per set of agitators is over 100,000. All cells are now provided with white cast-iron bottom liners about three-fourths of an inch thick. The substitution of these rigid, heavy-bottom liners for those of sheet iron or armorite has eliminated the trouble previously experienced from weaving and bulging, with consequent breakage of agitators.

The subaeration cells are provided with air inlets below the centers of the agitators through which air is introduced at a pressure of 1½ pounds per square inch. Two No. 3 low-pressure Root blowers are installed for this purpose. One of these, however, provides more than enough air for all cells, some air being continuously released by "blowoff."

TABLE CONCENTRATION

The small extent of gravity concentration practiced at this plant is indicated by the fact that 150 to 200 tons of final flotation tailings from each mill section per 24 hours are handled by three Deister Plat-O concentrating tables. These three tables are obviously inadequate to do efficient concentration work on this tonnage of tailings, which contain relatively large amounts of fine material. The primary purpose of these tables is to act as pilot machines and thereby to indicate the efficiency of flotation operations, the degree of oxidation of the ore, and the degree of grinding obtained. The tables produce small amounts of pyrite concentrates, which are added to the pyritic flotation concentrates. Occasionally appreciable amounts of oxidized lead minerals are obtained in the table concentrates, but the average recovery of lead from this source is almost negligible. The table tailings are the final reject products of the concentrator.

One additional Deister Plat-O table per mill section serves as a pilot machine in the zinc circuit. The concentrates produced from the first cell of the circuit are pumped to this table, where they are spread out for observation by the operator. The table does not produce separate products, but the appearance of the small galena and pyrite streaks at the upper border of the material as the feed spreads on the table indicate the lead and pyrite contents of the concentrates. The amount of galena that could be removed as separate table lead concentrates is negligible, the assays of table feed and table tailings after the removal of galena concentrates being practically the same.

All Deister tables operate at a speed of 290 strokes per minute. Their plateaus have been removed.

DEWATERING AND HANDLING OF CONCENTRATES

As previously stated the division of the concentrator into three sections ends with the production of the various concentrates. The like products of all sections are united and pumped to their respective thickeners by Wilfley pumps.

There are seven 50 by 10 foot Dorr thickeners in the thickener building, three of which, as indicated in the description of flotation practice, serve as conditioning tanks in the zinc circuits of the three mill sections. The remaining four serve as thickeners for concentrates, one tank each for the lead, zinc, and iron concentrates and one held in reserve if it is desired to keep a product of any mill section separate.

The four concentrates thickeners operate with 4-inch Dorrco duplex-diaphragm pumps, which are directly connected to the thickener underflows. The pumps are equipped with ball valves iron covered with heavy rubber. The Dorrco pumps deliver the thickened pulps to 2-inch Wilfley pumps, which in turn deliver the pulps to their respective filters. Each Wilfley-pump discharge line is equipped with a by-pass line and valve which allows the return of thickened pulp to the feed well of the thickener if desired. The thickness of concentrates pulp is controlled by this return of pulp to the thickener. When a pulp becomes too dilute for rapid filtering the by-pass valve is opened, causing the thickener-discharge pulp to return to the thickener. The circulation of pulp is maintained until the pulp reaches the desired content of solids. This method of pulp-density control eliminates the raising or lowering of rakes, and all rakes are kept at their lowest points, without tension showing.

The use of considerable amounts of spray water dilutes feed to the thickeners. The pulps contain 15 to 30 per cent of solids, the higher figure applying more to lead concentrates because of their higher specific gravity. The thickened concentrates contain 50 to 75 per cent of solids, and filtering is started at a figure approaching the higher solids content and discontinued at about the lower figure. The overflows of the lead and zinc concentrates thickeners pass to ponds which in turn produce waste overflows. The overflows of the pyrite thickener and the three conditioning thickeners are discarded without further settling.

There are four 6-foot-diameter American 10-sector disk filters, one for each kind of concentrates pulp and one to serve the auxiliary thickener. The lead and zinc filters are each equipped with five disks, the pyrite filter with eight and the auxiliary with six. When in operation each filter tank receives the full thickener discharge, and the quantity is maintained in sufficient volume so that a portion overflows and passes to the Wilfley pump which feeds the original flotation-concentrates pulp to the thickener. This method of operation insures a high pulp level in the filter tank at all times, thereby keeping all sectors immersed when in the low position which in turn prevents loss of suction.

During normal operation vacuum at the filters is between 21 and 23 inches of mercury and is maintained by two type ER-1 Ingersoll-Rand air compressors, one 22 by 8 inches and the other 18 by 6 inches in size. They are connected to receiving tanks, which in turn are connected to the suction lines of the filters. Air for blowing the sector bags at their discharge positions is furnished at 1½ to 2 pounds pressure by one 7 by 5 inch Ingersoll-Rand compressor.

The filter bags last 60 to 90 days and are made of rather coarse cotton-twill cloth. The filtering of pyrite is the most severe on the bags and that of zinc concentrates the least severe.

The cakes from the various filters drop to 100-ton-capacity steel bins directly beneath the filters. They have V-shaped bottoms and are equipped with sliding gates which are operated by ratchet-and-pinion worm-gear mechanisms. A spur railroad track runs directly under the concentrates bins, an arrangement that permits filter cakes to be dropped directly from the filters into cars spotted on this track.

The filtered concentrates usually have a moisture content as follows:

Concentrates	Per cent
Lead	8.0 to 8.5
Zinc	9.5 to 10.0
Pyrite	10.5 to 11.0

The various filtrates are sent to their respective ponds, which also receive the overflows from the concentrates thickeners. The overflows of the ponds are waste products; the ponds are cleaned about once every two years. The addition of alkaline reagents to promote settling and filtering or the heating of pulps to be thickened has not been practiced at this plant.

DISPOSAL OF TAILINGS

Approximately one-half of the total tonnage milled leaves the plant as waste tailings. The tailings pulp of each concentrator unit, after leaving the tables, flows by gravity to an electrically controlled automatic sampler which cuts the stream at right angles, at regular intervals and discharges the sample into a receptacle reserved for the purpose. The pulp stream then flows to the main tailings launder, where it joins the tailings pulps of the other two concentrator sections. The combined pulps leave the mill building, pass under the railroad yard tracks, and are conveyed by launders to one of two ponds. One pond

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is held in reserve for emergencies. These ponds are surrounded by embankments of old tailings and are constructed with weir overflows at various points for the discharge of clear water into the Jordan River. The combined area of the two ponds is approximately 60 acres.

WATER SUPPLY

Two 5-inch Fairbanks-Morse centrifugal pumps in a small pump house elevate water from concrete sumps, which are supplied by a canal, to a water tank at the mill tower, a vertical distance of about 100 feet. These pumps are controlled electrically by means of float switches at the water tank which start and stop the pumps automatically, thus assuring an ample supply of water at all times and without waste. Each pump is directly connected to a 35-hp. motor which operates at a speed of 1,800 r.p.m., and each is capable of pumping water at the rate of over 750 gallons per minute. As the concentrator uses approximately 1 gallon of water per pound of ore milled these pumps are kept quite busy at the present 1,000-ton plant capacity rate. As the supply of water for concentrator purposes is adequate return water is not introduced into the mill circuit.

The canal water has a pH value of between 7.6 and 7.8, which probably accounts for the fact that alkaline reagents such as soda ash are not required in the flotation circuits. The pH value of the water in the lead flotation circuit remains at about the figure given for new water, the introduction of the sulphite reagent having very little effect on this value. In the zinc circuit the pH value decreases slightly to about 7.6, owing to the addition of copper sulphate. The pyrite circuits have pH values of about 8.0, the increase being accounted for by the addition of the alkaline sodium sulphide reagent.

PUMPING

As the mill is built on flat ground more pumping of the various flotation products is required than would be needed with a hillside site. The pumps required are concentrated in three different locations within the mill. The seven 2-inch Wilfley pumps, one a spare, which handle classifier-overflow pulps are located in a concreted passageway below the rod-mill floor. Three of these pumps are used for elevating the rod-mill classifier-overflow pulps through a short lift to the ball-mill classifiers on the ground floor. The remaining three elevate the finished classifier overflow pulps through a vertical distance of about 50 feet to the lead-circuit surge tanks. By a system of pipe lines and flexible hose equipped with couplings one pump can be substituted for another with the least loss of time or material. In this passageway there are also three 2-inch American floor pumps for the return of spills to a classifier. These pumps are placed in concreted sumps and are equipped with float switches designed to start and stop them automatically. It is customary, however, to operate these floor pumps with push buttons rather than by the automatic devices provided, as the latter cause too much flooding of the lead flotation circuit.

Another concreted passageway for pumps is below the center of the ground floor of the main building. This passageway contains 19 Wilfley pumps which handle the various flotation concentrates, middlings, and tailings products. The three that handle the tailings pulps of the primary zinc machines are 3 inches in size, and all others are 2 inches. Curbs are built into the concrete floor of the passageway to prevent intermixing of spills of the different flotation circuits. Nine 2-inch American floor pumps, one for each curbed area, are provided to return lead-circuit spills to the respective lead-circuit surge tanks, zinc-circuit spills to the conditioning thickeners, and pyrite spills to the tailings launders.

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The thickener building contains seven 2-inch Wilfley pumps, one for each thickener. Three of these return conditioned zinc-circuit pulps to the secondary zinc flotation machines. The remaining four feed thickened pulps to the filters. This building is also equipped with seven automatic 2-inch American floor pumps placed in sumps at the bottom of concreted passageways and under the various thickeners. These pumps automatically return any spills to their respective thickeners.

The length of service obtained from the various pump runners and plates varies within wide ranges. Those of pumps operating on initial flotation feed possibly average two weeks. Those handling lead and zinc concentrates average several months, while those operating with pyritic concentrates give poor service, particularly the pump that feeds thickened pyritic concentrates to the filter; these parts average only about six days. The average continuous service of Wilfley pump runners and plates for the entire plant is about 30 to 35 days. The casings have a somewhat longer life.

HANDLING OF REAGENTS

All reagents, with the exception of copper sulphate, are made up into solutions or oil mixtures on the top floor of the main concentrator building. The reagents, contained in drums or barrels, are brought to this floor by an elevator of 2,500 pounds capacity operated with push-button control. Six hoppers are available for receiving the separate dry reagents, and if necessary lumps are broken up before the reagents are loaded into these hoppers. In making a solution a computed amount of the dry reagent is weighed out into a barrow placed upon floor scales. This charge is then dumped into one of four 250-gallon capacity reagent-mixing tanks equipped with Devereaux agitators. The agitators are kept free of salts by cylindrical screens which surround them. After the solution of the salt is obtained it is tapped into one of five 1,000-gallon-capacity supply tanks. It is Midvale practice to figure 20 pounds of dry reagent per inch of height of the supply tanks, and this is equivalent to a solution strength of about 13 per cent.

The individual oils used, which comprise Republic 19 B, Yarmor steam-distilled pine, and Hardwood Creosote No. 2, are introduced without mixing into three 250-gallon-capacity oil tanks. Republic 19 B and pine oil are pumped to these tanks from their respective 14,000-gallon-capacity supply tanks, which are located on the ground near the mill building. The Hardwood Creosote is tapped from drums into its tank. Three 1,000-gallon-capacity supply tanks are placed below the three 250-gallon oil tanks and oil mixtures are made up in these tanks.

The eight 1,000-gallon-capacity supply tanks, five for solutions and three for oil mixtures, are connected to the various Geary reagent feeders by ¼-inch pipe lines. A mixing tank is not required for xanthate, as the ready solubility of this reagent permits adding it directly to one of the 1,000-gallon-capacity supply tanks.

CONVEYING AND ELEVATING

As previously described, the crude ore is fed from two 75-ton-capacity receiving bins at the crushing plant upon two 36-inch continuous manganese-steel apron feeders, which travel toward each other at the rate of 8 feet per minute. These feeders discharge their loads at a level of 5 feet above the ground to a lined hopper which supplies a continuous chain elevator.

The chain elevator, which travels at a speed of 50 feet per minute, carries the ore at an angle of approximately 80° from the horizontal to a discharge point about 30 feet above the ground level. This elevation gives the necessary fall for the operation of the mechanical grizzly and the gyratory crusher.

The grizzly undersize and crusher-discharge products join upon an 18-inch belt conveyor about 4 feet above ground level. This conveyor, which is 17 feet long between centers, discharges to the first belt elevator. The latter, equipped with 16-inch buckets, raises the ore vertically to a discharge point about 50 feet above ground level, which provides the necessary fall for the initial screening and crushing operations. The first roll-discharge product is received by an 18-inch belt conveyor, 32 feet long between centers, at about ground level, and this conveyor feeds the second 16-inch bucket elevator.

This second elevator raises the ore vertically to a point about 50 feet above ground level, and this height provides sufficient fall for the final screening and crushing operations.

The final crushed material, representing the combined screen undersize products, is fed at a point about 3 feet above ground level upon an 18-inch inclined belt conveyor which is about 200 feet long, horizontally, between centers. The latter, which travels at a speed of 200 feet per minute, conveys the crushed ore to the bin-storage department of the main building, where it is discharged to the automatic chain sampler about 45 feet above ground level.

After sampling, the material passes to a horizontal 16-inch belt conveyor 150 feet long between centers, which travels above the center line of the row of storage bins at a speed of 300 feet per minute. This belt is equipped with an automatic tripper. The loaded portion of the belt is about 3 feet above the top of the bins and about 40 feet above ground level.

The discharge points of the various roll feeders at the bottoms of the bins are about 15 feet above the ground. From here the ore is fed to 16-inch reversible gathering conveyor belts, which in turn discharge to 12-inch conveyor belt feeders. These feeders discharge ore to the various grinding mills. All grinding mills and classifiers are at ground level, as indicated in Figure 2.

TABLES

Table 1 presents screen analyses of intermediate and final concentrator products. Table 2 gives metallurgical data covering the entire fourth quarter of 1929. Power distribution for November, 1929, is shown in Table 3, and the wage scale and a summary of labor in Table 4.

Table 1.- Screen analyses of intermediate and final concentrator products

Tyler screen sizes	Rod mill				Ball mill			Concentrates			Final tailings
	Feed	Discharge	Classifier sand return	Classifier overflow	Classifier sand return	Discharge	Classifier overflow	Lead	Zinc	Iron	
Minus 1 inch plus ½ inch	6.1	- -	- -	- -	- -	- -	- -	- -	-	- -	- -
Minus ½ inch plus 3 mesh	12.8	- -	- -	- -	-	- -	- -	- -	- -	- -	- -
Minus 3 mesh plus 4 mesh	19.2	- -	- -	- -	- -	- -	- -	- -	- -	- -	- -
Minus 4 mesh plus 6 mesh	6.6	- -	- -	- -	- -	- -	- -	- -	- -	- -	- -
Minus 6 mesh plus 8 mesh	4.7	- -	- -	- -	- -	- -	- -	-	- -	- -	- -
Minus 8 mesh plus 10 mesh	4.6	- -	- -	- -	- -	- -	- -	- -	- -	- -	- -
Minus 10 mesh plus 14 mesh ..	3.5	- -	- -	- -	- -	- -	- -	- -	- -	- -	- -
Minus 14 mesh plus 20 mesh ..	4.3	0.4	0.7	0.2	0.2	0.2	- -	- -	- -	- -	- -
Minus 20 mesh plus 28 mesh ..	4.5	1.4	1.7	.7	.7	.6	- -	- -	- -	- -	- -
Minus 28 mesh plus 35 mesh ..	5.3	7.4	12.0	3.5	3.7	.7	0.4	- -	- -	- -	1.0
Minus 35 mesh plus 48 mesh ..	4.3	16.3	24.5	7.7	8.6	3.2	1.3	- -	- -	- -	2.9
Minus 48 mesh plus 65 mesh ..	3.7	19.2	23.8	12.7	33.2	10.0	3.7	0.3	0.8	1.4	8.8
Minus 65 mesh plus 100 mesh..	3.7	15.0	17.5	16.0	20.8	30.5	9.1	.8	2.5	10.6	16.8
Minus 100 mesh plus 150 mesh	1.3	4.9	4.8	5.5	9.6	10.8	4.2	1.4	3.4	11.0	7.0
Minus 150 mesh plus 200 mesh	2.8	8.2	6.2	14.2	12.0	18.3	21.2	9.4	16.4	24.4	11.9
Minus 200 mesh	12.6	27.2	8.8	39.5	11.2	25.7	60.1	88.1	76.9	52.6	51.6
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 2.- Metallurgical data covering entire fourth quarter of 1929

Total ore treated	tons	88,353
Time operated	days	90
Operation per day	hours	24
Average number of sections operated		3
Average ore treated per 24 hours	tons	982
Average ore treated per section per 24 hours	do.	327
Total lead concentrates produced	do.	16,287
Average lead concentrates produced per 24 hours	do.	181
Total zinc concentrates produced	do.	9,805
Average zinc concentrates produced per 24 hours	do.	109
Total pyrite concentrate produced	do.	19,658
Average pyrite concentrate produced per 24 hours ..	do.	218
Total concentrates produced	do.	45,750
Average concentrates produced per 24 hours	do.	508
Ratios of concentration·		
Lead concentrates	tons into 1	5.42
Zinc concentrates	do.	9.00
Pyrite concentrates	do.	4.50
Total concentrates	do.	1.93

Table 3.- Power distribution for November, 1929

Total ore concentrated	tons	29,871
Lead concentrates produced	do.	5,567
Zinc concentrates produced	do.	3,084
Pyrite concentrates produced	do.	6,853

	Kw. h. per dry ton of ore milled	Per cent of total	Kw. h. per dry ton of concentrates produced
Crushing and conveying	1.51	5.57	- -
Grinding	9.15	33.79	- -
Flotation			
Lead circuit	2.60	9.59	14.0
Zinc circuit	4.78	17.64	46.5
Pyrite circuit	.86	3.17	3.8
Tables	.25	.92	- -
Thickeners	.48	1.77	- -
Filtering	.59	2.18	- -
Pumping	6.15	22.71	- -
Water supply	.72	3.66	- -
Total	27.09	100.00	52.4 kw. h. per ton of total concentrates produced

As it is obviously impossible to compare power consumed without reducing the figures to a comparative basis, the power consumed per ton of the various dry concentrates are given for the flotation operations. It is noted that in plants producing concentrates amounting to one-half the weight of the mill feed and furthermore when three different concentrates are produced, all by flotation methods, that considerable more flotation capacity and power are required than would be the case in a plant which produces but one concentrates product. This is a fact not usually noted in published statements regarding power consumed at various plants.

Table 4.- Wage scale and summary of labor for April, 1929

	Number of shifts employed				Rate	Total
Job	Day	Afternoon	Night	Total	per shift	amount
Concentrator						
General foreman	1	0	0	1	$6.75	$ 6.75
Shift foremen	1	1	1	3	5.90	17.70
Head repairman	1	0	0	1	5.10	5.10
Shift repairmen	1	1	1	3	4.60	13.80
Machinist	1	0	0	1	5.75	5.75
Machinist helper	1	0	0	1	3.70	3.70
Electrician	1	0	0	1	5.75	5.75
Flotation operators	3	3	3	9	4.75	43.75
Rod mills	1	1	1	3	4.40	13.20
Pumps	1	1	1	3	4.40	13.20
Filters	1	1	1	3	4.40	13.20
Thickeners	1	1	1	3	3.75	11.25
Feeders	1	1	1	3	3.25	9.75
Bins	2	1	2	5	3.25	16.25
Clean-up men	2	1	1	4	3.25	13.00
Tailings dike	2	0	0	2	3.25	6.50
Reagent man	1	0	0	1	4.40	4.40
Reagent helper	1	0	0	1	3.75	3.75
Concentrates loaders	4	0	0	4	3.25	13.00
Sampler	1	0	0	1	3.75	3.75
Roustabout	1	0	0	1	3.25	3.25
Watchman	0	½	½	1	3.40	3.40
Crushing plant						
Head man	1	0	0	1	4.75	4.75
Pan feeder	1	0	0	1	3.75	3.75
Screens	1	0	0	1	3.75	3.75
Rolls	1	0	0	1	4.00	4.00
Clean-up	1	0	0	1	3.25	3.25
Unloading ore	5	0	0	5	3.40	17.00
Total	39	12½	13½	65		$266.70
Winter months						
Firemen at mill				3	3.60	10.80
Firemen at thaw shed				3	3.40	10.20

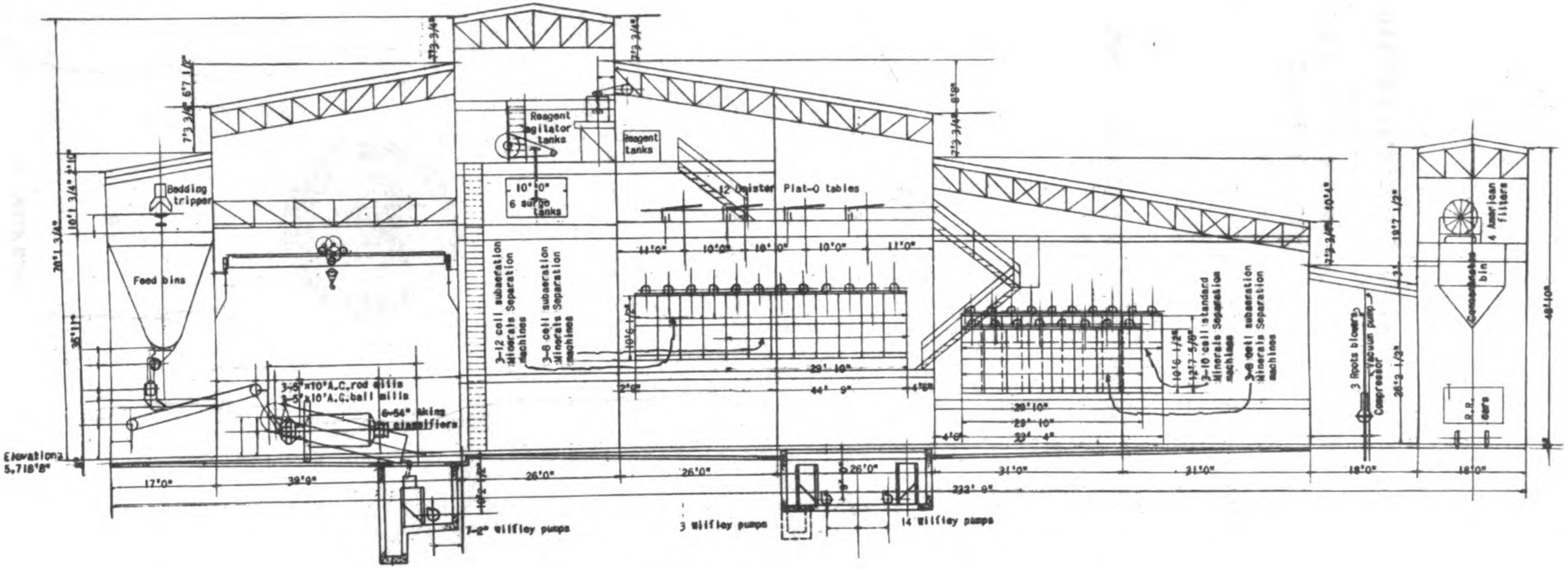

Figure 2.— Concentrator building, transverse section